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GRAVITY AND FLOTATION CONCENTRATION OF GOLD IN A PLACER SAND

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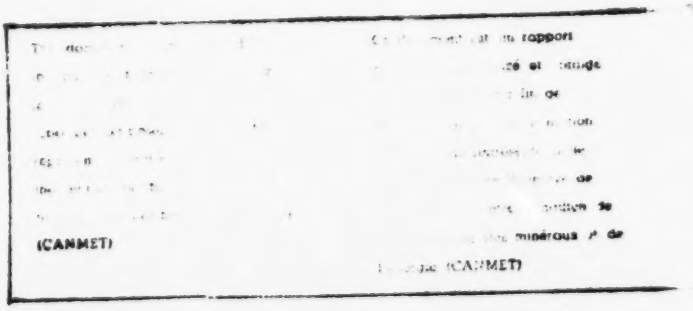

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GRAVITY AND FLOTATION CONCENTRATION OF GOLD IN A PLACER SAND

by

T.F. Berry*

SUMMARY

As is the case with most placer sands a careful survey must be made of the Saskatchewan River claims to ensure that low-grade sands are bypassed in the mining phase of the operation.

Preconcentration of the sand under investigation, by the screen removal of barren or uneconomic fractions, is a prerequisite to gold concentration.

The test results show that flotation concentration of the gold is an economically viable alternative to the presently used gravity technique.

Regardless of the concentration process used, whether by gravity or floation, an irreducible tailing ranging between 0.034 and 0.069 g/t (0.001 and 0.002 oz/ton) was obtained.

Concentrate grades and recoveries were generally high. A gravity concentrate assaying 146.260 g/t (4.266 oz/ton) with a recovery of 97.16% (Table 4) compares with a flotation concentrate assaying 213.938 g/t (6.240 oz/ton) and a recovery of 97.80% (Table 10).

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CONCENTRATION PAR GRAVITÉ ET PAR FLOTTATION DE L'OR D'UN SABLE ALLUVIAL

par

T.F. Berry*

RÉSUMÉ

Comme c'est le cas de la plupart des sables alluviaux, les claims de la rivière Saskatchewan doivent faire l'objet d'un levé détaillé pour garantir que, les sables à faible teneur ne seront pas extraits pendant l'exploitation.

Une première concentration des sables étudiés, effectuée par élimination au tamis des fractions stériles ou non rentables, est une condition préalable pour la concentration de l'or.

Les résultats des essais montrent que la concentration de l'or par flottation est une solution plus rentable que la technique par gravité utilisée actuellement.

Quelque soit le procédé de concentration utilisé, par gravité ou par flottation, on a obtenu des résidus irréductibles titrant entre 0,034 et 0,069 g/t (0,001 et 0,002 oz/tonne.

Les teneurs des concentrés et les taux de récupération ont été généralement élevés. Un concentré obtenu par gravité, titrant 146,260 g/t (4,266 oz/tonne) a permis une récupération de 97,16 % (tableau 4), et un concentré obtenu par flottation, titrant 213,938 g/t (6,240 oz/tonne), a permis une récupération de 97,80 % (tableau 10).

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INTRODUCTION

BACKGROUND

Gold has been recovered from placer sand deposits in Canada, most visibly in the Yukon Territory, for over one hundred years. This activity reached a zenith at the turn of the century with the fever generated by the Yukon "gold rush".

As early as 1870 the North Saskatchewan River, near North Battleford (1), was the scene of intensive operations and to this day gold is being recovered from the sand bars. Most of the operations on this river were small with the employment of dredges to mine the sand and sluice boxes to concentrate the gold.

Where only one valuable mineral is present e.g. gold, and the placer sands are simple accumulations of readily dispersed particles lending themselves to easy separation and concentration, the recovery processes tend to be simple (2). Indeed, there are some highly profitable operations requiring only the use of screens, pumps and sluice boxes to produce a finished product.

While the sluice box, which requires a large volume of water, will concentrate the coarser gold particles [+74 μm, (+200 mesh)], its efficiency decreases markedly in the concentration of the fine, flakey gold usually associated with mud, clay and slime fractions of the sand.

More sophisticated gravity concentration operations employ jigs, shaking tables, pinched sluices, Reichert cones, spirals and hydrocyclones. In Cornwall, England, fine cassiterite is concentrated on a round table. Even with the use of these controlled devices, the finest gold particles, which may be an appreciable percentage of the gold in the sand, will escape recovery.

The concentration of gold by froth flotation has been known since flotation was in its infancy. The low price of gold at the time mitigated against this method of recovery. It is only with the relatively recent rise in the price of gold to its present level that this recovery method becomes a viable alternative to gravity concentration.

OBJECT OF INVESTIGATION

In 1983, CANMET initiated a gold program, one facet of which involved a close look at the current "state of the art" as applied to placer deposits. Coincidentally Mr. W.A. Bonter, of Saskatchewan River Gold Company, visited CANMET and asked if we would start an investigation on sands from his placer claims on this river with the object of developing an alternative to his currently used sluice box concentration procedure.

ORE SAMPLES

In September 1983, four drums of wet placer sand, designated sluice-box feed, were received. At the same time a small amount (2500 grams) of sluice-box concentrate, referred to as a "black sand", was received.

In September 1984, an additional shipment of sluice-box feed weighing 1111.313 kg (2450 lb) arrived at CANMET to provide sufficient material for a continuous process.

MINERALOGY*

The results and conclusions of a mineralogical examination on a sample of rougher magnetics obtained during the processing of the black sand sluice box concentrate were as follows:

"General Mineralogy: The heavy minerals found during the mineralogical examination consisted mainly of magnetite, hematite and ilmenite with lesser amounts of various silicates. A few grains of chromite and of cassiterite were also observed. The following sulfides, in decreasing order of abundance were found as small inclusions mainly in magnetite: pyrite, chalcopyrite, pyrrhotite, and arsenopyrite. Free grains of arsenopyrite were also found as well as grains of a Fe-As mineral, possibly loellingite. Numerous grains of iron were also observed. Native gold was the most common mineral of economic interest to be found in this sample. A total of 275 grains were found in these polished sections followed by Pt-Fe alloy grains (78 grains found), and some containing various small inclusions of other platinum-group minerals (PGM). One liberated grain of sperrylite ($PtAs_2$) was also observed.

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*Internal Report M-3408, Mineralogy Section, Mineral Sciences Laboratory, CANMET, Energy, Mines and Resources Canada.

Detailed Mineralogy:

i) Gold: Particles of gold were found in all the various non-magnetic fractions, increasing in number [8 grains found in +295 µm, (+48 m), 171 in -147 + 74 µm (171 in -100 + 200 m)] with decrease in mesh-size. Few particles were found in the magnetic fractions, but these particles were carried in the magnetic fractions, being trapped in clusters of strongly magnetic minerals, e.g. magnetite, while separating magnetic from non-magnetic minerals with the hand magnet.

All the gold particles observed were liberated, normally flattened grains or flakes, being lenticular in polished sections. In the larger fractions [for example in the +295 µm (48 m)] the large gold particles are really clusters of smaller particles attached together. The gold particles generally show zoning caused by a variation in composition generally this core is silver-enriched, with a nearly pure gold rim.

The presence of mercury was found in many gold grains in this sample. An explanation for this would be the extensive use of mercury in the old days by prospectors, who were using mercury to recover fine gold particles, which then contaminated the river beds. This mercury dumped into the river, readily amalgamated with the available gold particles, which were collected later on ....

ii) PGM: A total of 78 Pt-Fe alloy grains were found, in this sample, together with one grain of sperrylite ($PtAs_2$). All were liberated grains with exception of two Pt-Fe grains, one which had minor quartz attached, the other had minor calcite. These grains were concentrated in the -147 + 74 µm (-100 + 200 m) fractions, as 96% of the grains observed were found in that fraction and they varied in size, between 10 x 60 µm to 20 x 275 µm (in cross-section). They were quite irregular in shape but generally flattened as observed in cross-sections. Although quite homogeneous in composition, a lot of these Pt-Fe alloy grains contained various small PGM inclusions which were semi-qualitatively analysed by the energy dispersive system of the scanning electron microscope. Examples are: four grains in which small inclusions of cooperite (PtS) or braggite (Pt, Pd) S were observed; three grains in which inclusions of an undetermined Rh-S mineral were found, two grains where iridosmine, (Os, Ir, Ru) was observed and one grain where the presence of irarsite (IrAsS) or hollingworthite (RhAsS) was found.

Full quantitative analyses using the electron probe microanalyser were done on 18 of these Pt-Fe alloy grains. According to their compositions, one of these is native platinum, while the others are all very close to Pt_3Fe (isoferroplatinum). Replacement of Pt by Pd, Rh, Ir or Os and of Fe by Cu and Ni is quite common in these alloys.

There is no evident explanation to the fact that so many Pt-Fe alloy grains (41%) were found in the magnetic fractions. The most plausible explanation would be that the grains found in the magnetic fractions were trapped in clusters of strongly magnetic minerals, e.g. magnetite, while separating magnetic from non-magnetic minerals with the hand magnet, as mentioned earlier for the gold particles. Varying magnetic susceptibility of these Pt-Fe alloy grains would also be responsible for this.

Conclusions:

1. All the gold particles found in this sample were liberated grains.
2. All the Pt-Fe alloy grains with exception of two, and the single sperrylite grain were free of silicates and oxides.
3. All the other PGM present; undetermined Rh-S mineral, iridosmine, braggite/cooperite, irarsite/hollingworthite occurr as intergrowths within the Pt-Fe alloy grains.
4. 96% of the Pt-Fe alloy grains were found in the -174 + 74 µm (-100 + 200 m) fraction."

SCOPE

All placer sands contain the whole range of sizes from coarse to fine. A screen analysis was done to determine to what point preconcentration should proceed for the removal of uneconomic fractions. The remaining fractions with a greatly enhanced assay ocassioned by the removal of the coarse material were subjected to gravity concentration (jigs and tables), to magnetic separation and to flotation.

The sluice-box concentrate was treated separately by magnetic separation and table concentration.

DISCUSSION AND EVALUATION OF TEST PROCEDURES AND RESULTS

COMPARISON OF TWO SAND SHIPMENTS

The two shipments of placer sand, although received a year apart, were assumed to have been mined from the same location and would be nearly identical insofar as the gold distribution was concerned. This was an erroneous assumption as the following results show. Each shipment was wet screened on a 4699 µm (4 mesh) screen and then re-screened to remove the +589 µm (+30 mesh) fractions. The selection of the 589 µm screen will become apparent with a perusal of the results of a screen analysis shown in Table 3.

Table 1 - Screen results on shipment No. 1

Product		Weight	Assay		Distribution
µm	mesh	%	g/t	oz/ton	Au, %
+4699	4	43.39	barren		-
+589	30	22.68	0.034	(0.001)	0.64
-589	30	33.93	3.120	0.091	99.36
Total (calcd)		100.00	1.063	0.031	100.00

Table 2 - Screen results on shipment No. 2

Product		Weight	Assay		Distribution
µm	mesh	%	g/t	oz/ton	Au, %
+4699	4	35.05	barren		-
+589	30	29.68	0.069	0.002	17.14
-589	30	32.27	0.309	0.009	82.86
Total (calcd)		100.00	0.137	0.004	100.00

The lowest detectable level in our assaying procedure is 0.034 g/t (0.001 oz/ton). The 4699 µm fractions in Tables 1 and 2 assayed <0.001 oz Au/ton. Since the actual assay was uncertain and in any case was uneconomic from a mineral dressing point, it was decided to designate these fractions as barren.

Table 3 - Screen analysis on sand shipment No. 1

Product		Weight	Assay, Au		Distribution
µm	Mesh	%	g/t	oz/ton	Au, %
+2362	8	17.04	0.171	0.005	2.00
+1651	10	6.94	0.069	0.002	0.33
+1168	14	5.03	0.103	0.003	0.35
+589	28(1)	9.20	0.034	0.001	0.21
+417	35	6.25	3.566	0.104	15.32
+219	48	18.40	0.686	0.020	8.67
+208	65	17.95	0.789	0.023	9.73
+147	100	12.55	1.163	0.066	19.51
+104	150	3.44	4.971	0.145	11.76
+74	200	1.75	19.062	0.556	22.93
+53	270				
+44	325	1.44	9.291	0.271	9.19
-44	325				
Total (calculated)		100.00	1.440	0.042	

(1): 28 mesh in the Tyler series is equivalent to 30 mesh in the A.S.T.M. series

Apart from the sand in shipment No. 2 being of lower grade (0.137 g/t vs 1.063 g/t), 17.14% of the gold in this shipment was lodged in the -4699 + 589 µm fraction compared to 0.64% in the same fraction in the first sand shipment.

As with most gold-bearing placer sands the gold is seldom evenly distributed. Horizons will range in assay from barren to low and high grade and this circumstance then presents a mining problem. It is essential that the orebody be surveyed and sampled to ensure that zones which are barren or contain uneconomic amounts of gold can be bypassed in the mining process.

If the sands persist in such an uneven gold distribution it may be necessary to install a high capacity jig to handle the -4699 µm material as a safety valve. The jig tailing would then be screened to remove the +589 µm fraction and the -589 µm fraction would proceed to the concentration process.

SCREEN ANALYSIS

Only one screen analysis was done during the investigation, that being on sand shipment No. 1. Samples riffled from this sand comprised the feed for all the bench-scale concentration tests. The object of the screen analysis was to determine which fractions, if any, of the -4699 µm material might be discarded as barren or as being uneconomic. These results may be seen in Table 3.

The +589 µm fractions when combined, assayed a calculated 0.110 g/t (0.0032 oz/ton) and contained 2.89% of the gold. This material represented 38.21% of the -4699 µm sand and should be removed prior to any concentration process as uneconomic. The resulting feed to a concentration process, all -589 µm, occassioned by the removal of this large fraction, will assay a calculated 2.297 g/t (0.067 oz/ton) and will contain 97.11% of the gold.

PRECONCENTRATION

Using the simple and relatively inexpensive expedient of removing by screening a large barren or uneconomic fraction, the grade of the sand feed to a concentration process can be greatly enhanced.

In the case of shipment No. 1, the sand as received, which was assumed to be representative of that mined, assayed a calculated 1.063 g/t. With the removal of 43.39% of the material (the barren +4699 µm sand), and the subsequent removal of an additional 38.21% of the remaining sand (+589 µm) for a total discarded tailing of 65.02% of the sand received, the resulting feed to a concentration process (all -589 µm) was upgraded to 2.297 g/t (0.067 oz/ton) with a loss of only 2.89% of the gold.

The advantages of such a preconcentration step are obvious. Apart from enabling an operation to increase the plant throughput, the higher grade feed will result in a higher grade concentrate at an increased gold recovery. (This benefit may be seen in Table 15.)

GRAVITY CONCENTRATION

All gravity concentration devices are simply concerned with the recovery of heavy and the rejection of light minerals. Most placer gold is associated with magnetite and sometimes ilmenite which concentrates as a heavy mineral with the gold.

In a small operation, the appeal of the sluice box is in its simplicity of construction and operation. There are however certain disadvantages to its use. Access to a large body of water is necessary. This is usually a river to which the tailings are returned. Depending on the amount of fine clay in the mined sands, these rivers can become polluted. The sluice box is a labour-intensive intermittent operation in which the flow must be stopped or diverted to allow for concentrate recovery. Fine gold [-74 µm, (-200 mesh)] is often lost in the tailings as the magnetite packs the riffles rendering them ineffective as a concentrating device. A large weight of concentrate is usually recovered ensuring a re-concentration step as a necessary prelude to refining. Sluice box concentrates from placer sand usually contain a large amount of "black sand", basically magnetite which is best removed by a low-intensity magnetic separator. An additional loss of gold will result from this operation due to mechanical entrainment.

Three gravity concentration tests were done on large samples riffled from the No. 1 shipment of sand. In each of these tests the +589 µm (+30 mesh) fraction was removed and the -589 µm (-30 mesh) undersize was concentrated. The first of these tests involved jigging and tabling and the second and third test involved only tabling. The gravity concentrates were upgraded using a low intensity dry magnetic separator.

In Table 4, 38.55% of the feed was removed as a +589 µm oversize with a loss of 1.41% of the gold. The resultant feed to the concentration process (jig) assayed 3.086 g/t (0.090 oz/ton) containing 98.59% of the gold. The two non-magnetic fractions had a combined assay of 221.138 g/t (6.45 oz/ton) with a recovery of 94.13% of the gold.

Many placer sands contain platinum as well as gold. In two table concentration tests the results of which follow, the table cleaner concentrates were upgraded using a low intensity magnetic separator. Each of the products was assayed for platinum to determine the distribution of that element.

In Table 5, the table cleaner concentrate assayed 115.060 g/t (3.356 oz/ton) with a recovery of 90.85% of the gold. The same concentrate in Table 6 shows a grade of 163.574 g/t (4.771 oz/ton) at a recovery of 94.51%. The difference in grade and recovery can be related to the grade of the -589 µm feed [1.954 vs 2.229 g/t (0.057 vs 0.065 oz/ton)].

Table 4 - Test 1. Results of jigging, ro-cl tabling, magnetic separation

Product	Weight	Assay				Analysis %	Distribution %	
		Au		Pt				
	%	g/t	oz/ton	g/t	oz/ton	Sol Fe	Au	Sol Fe
Jig-non-mags	0.48	249.994	8.75	-	-	5.69	74.73	1.39
Jig mags	0.30	13.371	0.39	3.086	0.09	50.78	2.14	7.74
Jig conc (calcd)	0.78	189.939	5.54	-	-	23.02	76.87	9.13
Table non-mags	0.34	109.712	3.20	2.057	0.06	9.12	19.40	1.57
Table mags	0.16	10.628	0.31	4.457	0.13	44.30	0.89	3.60
Table cl conc (calcd)	0.50	78.170	2.28	-	-	20.38	20.29	5.17
Table cl tailing	4.18	0.171	0.005	-	-	1.23	0.36	2.61
Table ro conc (calcd)	4.68	8.503	0.248	-	-	3.28	20.65	7.78
Table ro tailing	55.99	0.034	0.001	-	-	1.11	1.07	31.58
Jig tail (calcd)	60.67	0.686	0.020	-	-	1.28	21.72	39.36
-589 µm feed (calcd)	61.45	3.086	0.090	-	-	1.62	98.59	48.49
+589 µm tailing	38.55	0.069	0.002	-	-	2.63	1.41	51.51
Head (calcd)	100.00	1.989	0.058	-	-	1.97	100.00	100.00

Table 5 - Test 2. Results of ro-cl tabling, magnetic separation

Product	Weight	Assay				Analysis %	Distribution %		
		Au		Pt					
	%	g/t	oz/ton	g/t	oz/ton	Sol Fe	Au	Pt	Sol Fe
Table non-mags	0.56	193.367	5.640	2.606	0.076	8.77	90.00	4.03	2.40
Table mags	0.39	2.640	0.077	2.914	0.086	51.15	0.85	3.10	9.75
Table cl conc (calcd)	0.95	115.060	3.356	2.743	0.080	26.17	90.85	7.13	12.15
Table cl tailing	5.79	1.029	0.030	<0.343	<0.010	1.42	4.96	5.44	4.01
Table ro conc (calcd)	6.74	17.108	0.499	0.686	0.020	4.91	95.81	12.57	16.16
Table ro tailing	53.83	0.069	0.002	<0.343	<0.010	1.10	3.08	50.47	28.93
-589 µm feed (calcd)	60.57	1.954	0.057	0.377	0.011	1.52	98.89	63.04	45.09
+589 µm tailing	39.43	0.034	0.001	<0.343	<0.010	2.58	1.11	36.96	54.91
Head (calcd)	100.00	1.200	0.035	0.377	0.011	2.05	100.00	100.00	100.00

Table 6 - Test 3. Results of ro-cl tabling, magnetic separation

Product	Weight	Assay				Analysis %	Distribution %		
		Au		Pt					
	%	g/t	oz/ton	g/t	oz/ton	Sol Fe	Au	Pt	Sol Fe
Table non-mags	0.42	252.338	7.360	2.194	0.064	11.11	78.51	2.58	2.37
Table mags	0.36	59.999	1.750	2.640	0.077	52.94	16.00	2.67	9.69
Table cl conc (calcd)	0.78	163.574	4.771	2.434	0.071	30.42	94.51	5.25	12.06
Table cl tailing	2.61	0.309	0.009	<0.343	<0.010	2.29	0.58	2.48	3.04
Table ro conc (calcd)	3.39	37.851	1.104	0.823	0.024	8.67	95.09	7.73	15.10
Table ro tailing	56.00	0.069	0.002	<0.343	<0.010	1.01	2.58	53.49	28.75
-589 µm feed (calcd)	59.39	2.229	0.065	0.377	0.011	1.45	97.94	61.22	43.85
+589 µm tailing	40.61	0.069	0.002	<0.343	<0.010	2.72	2.06	38.78	56.15
Head (calcd)	100.00	1.337	0.039	0.343	0.0100	1.97	100.00	100.00	100.00

The upgrading of the table cleaner concentrate in each of these tests was done on a Stearns dry magnetic separator set at 1 ampere. The feed rate to this equipment was doubled in the second of these tests and the results showing 16.00% of the gold in the magnetic fraction is directly attributable to mechanical entrainment. In the opinion of the author, wet, low intensity magnetic separation using a Dings, Stearns, Sala or Eriez machine would result in a lower gold loss in the magnetic fraction. We were unable to test this theory because of insufficient material.

UPGRADING A SLUICE BOX CONCENTRATE

One half of the sluice box concentrate (approx. 1200 grams) was tabled with the following results.

Table 7 - Results of tabling a sluice box concentrate

	Wt	Assay, Au		Distribution
Product	%	g/t	oz/ton	Au, %
Table concentrate	12.52	4035.002	117.69	74.29
Table tailing	87.48	199.882	5.83	25.71
Head (calcd)	100.00	679.872	19.83	100.00

These results illustrate how inefficient is the tabling of a black sand concentrate prior to the removal of the magnetics. The large percentage of the gold in the table tailing is attributable to the magnetite packing behind the table riffles thus rendering them ineffective as a concentrating device for the fine gold particles.

In the second test on the sluice box concentrate the magnetics were first removed by low intensity dry magnetic separation. Here a rougher magnetic fraction was removed (at 1 ampere) and the non-magnetic fraction repassed on the same machine (at 2 amperes).

While we were unable to further upgrade the cleaner non-magnetic fraction by tabling, because of insufficient material, we believe that with the removal of the large rougher and cleaner magnetic fractions (81.17% of the weight) such an operation would result in an extremely high grade concentrate with a high recovery.

Table 8 - Results of magnetic separation of sluice box concentrate

Product	Wt %	Assay, Au g/t	Assay, Au oz/ton	Distribution Au, %
Cleaner non-mags	18.83	3462.785	101.00	96.03
Cleaner mags	17.31	46.628	1.36	1.18
Rougher non-mags (calcd)	36.14	1826.362	53.27	97.21
Rougher mags	63.86	29.485	0.86	2.79
Head (calcd)	100.00	678.843	19.80	100.00

FLOTATION CONCENTRATION

A preliminary flotation test gave the encouraging results shown in Table 10. A second flotation test was done on a larger sample from shipment No. 1 to verify these results. In each of these flotation tests the reagent scheme and flotation conditions were identical except that the conditioning time for the Aerofloat 25 was reduced from 5 to 1 minutes. The combined rougher and scavenger concentrates were cleaned once in Test 2, Table 11.

Three 5000-gram samples were wet screened at 589 µm and the -589 µm fractions were floated separately using the reagents shown in Table 9. All rougher and scavenger concentrates were combined for cleaning.

In both of the flotation tests froth loading was extremely light due to a scarcity of hydrophobic particles. In spite of this, rougher-scavenger concentrate recoveries were very high with concentration ratios of 123:1 and 141:1. The cleaner operation was not very successful, as witness the rejection of 17.4% of the gold in the cleaner tailings in Table 11 - Test 5.

Table 9 - Flotation conditions Test 4

Operation	Time min	Solids %	pH	g/tonne (2) AF 208	Aero 301	AF 25	Remarks
Conditioning	5	38.2	7.5(1)	37	37		(1) Au floats best at pH between 6.8 - 7.8. pH adjusted using 6% H_2SO_4.
"	5					25	
Ro flotation	5						
Conditioning	5		7.8	37	37		
"	5					15	
Scav. flotation	5						

(2) Aerofloat 208 promoter - 0.15 lb/ton
Aero 301 xanthate - 0.15 lb/ton
Aerofloat 25 promotor - 0.10 lb/ton

Table 10 - Results of preliminary flotation Test 4

Product	Wt %	Assay, Au g/t	Assay, Au oz/ton	Distribution Au, %
Rougher concentrate	0.27	497.818	14.52	87.7
Scavenger concentrate	0.43	35.656	1.04	10.1
Combined concentrate (calcd)	0.70	213.938	6.24	97.8
Scavenger tailing	59.51	0.034	0.001	1.3
-589 µm feed (calcd)	60.21	2.400	0.070	99.1
+589 µm tailing	39.79	<0.034	<0.001(1)	0.9
Head (calcd)	100.00	1.543	0.045	100.00

(1) Assume 0.001 for calculation.

Table 11 - Results of flotation Test 5

Product	Wt %	Assay, Au g/t	Assay, Au oz/ton	Distribution Au, %
Cleaner concentrate	0.32	321.593	9.380	78.8
Cleaner tailing	0.49	46.285	1.350	17.4
Ro + Scav. conc (calcd)	0.81	155.071	4.523	96.2
Rougher tailing No. 1	20.81	0.069	0.002	1.1
Rougher tailing No. 2	18.97	0.034	0.001	0.5
Rougher tailing No. 3	19.46	0.034	0.001	0.5
-589 µm feed (calcd)	60.05	2.126	0.062	98.3
+589 µm tailing No. 1	13.60	0.069	0.002	0.7
+589 µm tailing No. 2	13.15	0.069	0.002	0.7
+589 µm tailing No. 3	13.20	0.034	0.001	0.3
Head (calcd)	100.00	1.303	0.038	100.0

Table 12 - Consolidated results Test 5

Product	Wt %	Assay, Au g/t	Assay, Au oz/ton	Distribution Au, %
Cleaner concentrate	0.32	321.593	9.380	78.8
Cleaner tailing	0.49	46.285	1.350	17.4
Ro + Scav. conc (calcd)	0.81	155.071	4.523	96.2
Rougher tailing	59.24	0.034	0.001	2.1
-589 µm feed (calcd)	60.05	2.126	0.062	98.3
+589 µm tailing	39.95	0.069	0.002	1.7
Head (calcd)	100.00	1.303	0.038	100.0

A second shipment of sand was received, which with the remainder of the first shipment provided sufficient material for a small scale pilot-plant test. The +4699 µm and the +589 µm fractions were removed leaving a -589 µm fraction for flotation (Tables 1 and 2).

The Denver No. 5 flotation cells which were installed for this test sanded out in a few minutes removing any possibility of a continuous mill run. This would not present any problem in a full plant operation as flotation cells have been designed and are in use on this relatively (-589 µm) coarse feed.

As a consequence, the entire -589 µm combined sand fraction, was batch floated in lots of 15 kg each at 38.5% solids in a single Agitair flotation cell. The rougher and scavenger concentrates were combined and samples of each flotation tailing were removed and combined for a single tailing assay.

Table 13 - Flotation conditions Test 6

	Time	Solids	pH	g/tonne		
				AF 208	(1) Z-6	AF 25
	min	%				
Conditioning	5	38.5	7.5	37	37	
"	2					25
Ro flotation	5					
Conditioning	5		7.8	37	37	
"	2					15
Scav. flotation	5					

(1) American Cyanamid no longer makes Aero 301 Xanthate.
Dow Chemical, Z-6 (potassium amyl xanthate) was substituted.

Aerofloat 208 promoter - 0.15 lb/ton
Z-6 xanthate - 0.15 lb/ton
Aerofloat 25 promoter - 0.10 lb/ton

Table 14 - Results of flotation Test 6 (Mill run)

Product	Wt %	Assay, Au g/t	Assay, Au oz/ton	Distribution Au, %
Table concentrate	0.01	1734.821	50.60	48.42
Table sand tailing	0.16	44.228	1.29	19.71
Table slime tailing	0.28	24.342	0.71	19.04
Flotation conc (calcd)	0.45	69.393	2.024	87.17
Flotation tailing(1)	32.45	0.069	0.002	6.22
-30 mesh feed (calcd)	32.90	1.029	0.030	93.39
-4 + 30 mesh, shipment No. 1	8.59	0.034	<0.001	0.77
-4 + 30 mesh, shipment No. 2	18.43	0.069	0.002	3.54
-4 + 30 mesh (discarded)	27.02	0.069	0.002	4.31
+4 mesh, shipment No. 1	16.45	-	barren	-
+4 mesh, shipment No. 2	23.63	0.034	(2)<0.001	2.30
+4 mesh, (discarded)	40.08	0.034	0.001	2.30
Combined shipments (calcd)	100.00	0.480	0.014	100.00

(1) by difference

(2) for calculation use 0.001

Immediately apparent in this test is the reduced grade and recovery in the flotation concentrate. This is a direct consequence of floating a low grade sand. The feed to flotation assayed 1.029 g/t (0.030 oz/ton) which is about half the assay of the feed in previous flotation tests. Table 15, which follows, amplifies this conclusion.

One of the objectives of this test was to obtain sufficient flotation concentrate to attempt an upgrading operation using table concentration. As the results in Table 14 show, this was not successful. Bearing in mind that

the feed to the laboratory Deister table was a flotation concentrate containing residual reagents, (however well-washed), the gold in the slime tailing can be attributed at least in part to table flotation.

The physical dimensions of a laboratory Deister concentrating table are small. In spite of a careful control of the feed rate, wash water volume, table inclination and table speed the residence time of gold particles on the table was very short. As the sand packed behind the riffles the extremely fine, flakey gold grains simply washed over the riffles into a sand and slime fraction.

Use of a full size table rated at about 1 tonne/h in plant practice and reduction of the feed rate by a factor of 8 should correct this problem. In England, extremely fine cassiterite is recovered on a Holman table with a wave-type deck at a feed rate of about 136 kg/h.

Table 15 - Relationship between feed and concentrate grades and recoveries

			Assays, Au, g/t[3]					
	Table	Wt	Feed to[1]		Final		Recovery	(2)
Operation	No.	%	preconc	conc	concentrate	tailings	Au, %	C/R
	4	1.28	1.920	3.086	146.260	0.034	97.16	78:1
Gravity	6	0.78	1.337	2.229	163.574	0.069	94.51	128:1
Concentration	5	0.95	1.200	1.954	115.060	0.069	90.85	105:1
Flotation	10	0.70	1.543	2.400	213.938	0.034	97.80	142:1
Concentration	11	0.81	1.303	2.126	155.071	0.034	96.20	123:1
	14	0.45	0.480	1.029	69.393	0.069	87.17	222:1

[1] preconc - preconcentration
conc - concentration

[2] C/R - concentration ratio: tons of feed required to produce one ton concentrate

[3] Si units only. To obtain Au, oz/ton divide g/t by 34.285.

CONCLUSIONS

Regardless of the concentration process used (gravity or flotation) an irreducible tailing assaying between 0.034 and 0.069 g/t (0.001 - 0.002 oz/ton) was obtained. To pursue this gold would be uneconomic as it would entail an almost doubling of the residence time to recover perhaps 50% of the tailing gold.

The results shown in Table 15 highlight the importance of preconcentration. In the case of the ore under investigation the removal of about 65% of the sand fractions as barren or uneconomic greatly enhanced the grade of the feed to the subsequent concentration process. This was accomplished using a relatively inexpensive screening procedure. With the low, discardable tailing produced during gravity and flotation concentration, this preliminary ore dressing technique automatically resulted in a high final concentrate grade and recovery.

It is essential that the sand claims be carefully surveyed and sampled so that low grade zones may be bypassed during dredging operations. The results shown in Table 14 illustrate to what degree a low grade feed to flotation concentration will influence the grade and recovery in a final concentrate.

Gravity operations will recover most or all heavy minerals. The resulting concentrate will contain a large amount of magnetite which can be removed by wet or dry low intensity magnetic separation. Some loss of gold will occur during this operation. At no time was there a sufficient amount of final concentrate to exhaustively pursue an upgrading procedure.

The flotation process rejected almost all of these iron minerals in the tailing, resulting in final concentrate grades and recoveries as high as those obtained in gravity concentration. The rejection of the "black sand" in flotation resulted in higher concentration ratios than was the case with gravity concentration.

Froth flotation is an obvious alternative or supplement to gravity concentration processes. These processes being relatively inexpensive, will remain the basic method for the recovery of coarse placer gold. However the characteristics of some placer gold, including fine sizes, flake-like particle shapes, porosity and hydrophobic surfaces, make extremely difficult the recovery of fine [-74 μm (-200 mesh)] particles using existing gravity concentration techniques.

The gold in the North Saskatchewan River, not unlike most gold in alluvial sands, is unevenly distributed, (Tables 1 and 2). Such a distribution dictates the use of a high capacity jig to remove the coarse gold from a -4699 μm feed. This would be followed by a screeen to remove the +589 μm fraction from a jig tailing and thence to flotation concentration for the -589 μm fraction.

Flotation processing of gold-bearing placer sands may require the recycling of process waters. This can be achieved by the use of high capacity thickeners of the inclined plate or lamella-type which are efficient, compact, portable solid-liquid separators. Such a recycling would reduce reagent costs and minimize environmental damage.

The operating cost of flotation is generally considered to be about $4.50/ton of feed. Simply put, this means that with a flotation feed assaying 3.43 g/t (0.01 oz/ton) and with a 90% recovery rate the operation is profitable with gold at $500.00/oz.

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